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ELEMENTS OF PLANNING IN DEEP MINING

D. F. COATES AND M. DICKHOUT

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STOPE-AND-PILLAR DESIGN FOR THE ELLIOT LAKE URANIUM MINES

D. G. F. HEDLEY AND F. GRANT

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Elements of planning in deep mining[1]

By D. F. COATES[2] and M. DICKHOUT[3]

■ Ground control is one of the major problems in deep mining because the stresses are great enough to cause sloughing of development openings, not to mention walls and faces in stopes, even in hard rock mines. At the same time, "deep" conditions can exist at modest depths either where tectonic stresses are equal to those due to gravity at depth or where the formations are weak.

For typical stope geometry, the passage of stress around the opening produces concentrations in the face and relaxation in the walls. Therefore rockbursting, as well as less violent forms of instability, usually occurs either in the face rock due to the high stresses or in the wall rock due to the fracturing consequent on release of stress. Variability in strength properties is not peculiar to rock, it is exhibited by all structural materials; consequently, failure can never be predicted with certainty — the concept of probability of failure must be adopted.

The essence of planning in deep mining is to avoid unnecessary, excessive stress concentrations. When an orebody is mined out, the geometrical conditions will be most crucial with regard to stress concentrations. However, without adequate planning it is quite possible to produce more severe stress conditions at intermediate stages with the attendant reduction in safety and increase in costs. It is essential to follow the principle of keeping stope faces in a longwall configuration. In addition, between adjacent development openings and between an opening and a major geologic structure appropriate pillars are to be provided and acute angle intersections avoided. The creation of pillars of any type must be examined very carefully for the magnitude of stress concentrations that they will produce; even in the case of shaft pillars, it may be preferable to remove the ore from the shaft pillar before the shaft is sunk.

The essential elements of mining are the extracting of ore from the earth's crust and the maintaining of adequate stability in the surrounding ground.

The ratio of the maximum stress in the formation to the strength of the rock can be used to predict critical stability conditions. "Deep" is often interpreted as meaning a depth of 5,000 ft or more, where stresses are high enough to cause sloughing of development openings, not to mention walls and faces in stopes. However, tectonic horizontal stresses are greater in many areas than gravitational stresses, hence "deep" conditions can exist at lesser depths; indeed, severe rock bursting caused the closure of a mine in Canada operating at depths of 500-700 ft. Also, rocks with low strength will produce "deep" failure patterns at modest depths, e.g. in Saskatchewan potash mines.

Clearly, one of the major problems in deep mining is the achieving of effective ground control and, more particularly, the minimizing of the risk of rockbursts.

Rock failure

Stress

Field stress conditions in rock formations are known to have many patterns. Gravitational loading alone results in the maximum stress being vertical with the horizontal stress being some fraction, like ⅓, of the maximum stress; this can be expected in weak, or incompetent rocks. In yielding rocks, like the various types of salt, indirect evidence suggests that the horizontal stress is equal to the vertical, gravitational stress.

On the other hand, tectonic action can produce horizontal stresses that are many times the vertical stress, and possibly of more importance is the observation that both horizontal and vertical stresses can change by 50% or more over distances of only a few hundred feet. Considering the origin and subsequent history of typical hard rock formations, such patterns are not surprising.

When openings are excavated in rock, the normal field stress, S_n, must pass around the opening and is concentrated in the abutments and in the pillars as shown in Fig. 1(a) and 1(b). The pillar stresses, being primarily dependent on the extraction ratio, can be very high, and the volume of ground subjected to these high stresses can be much greater (and hence more critical with respect to rockbursts) than for abutments. The transverse field stress, S_t, (acting in the horizontal direction for horizontal seams) produces relatively low stress concentrations in the walls, owing to the slot-like geometry of the opening. However, if bedding, schistocity or laminations exist, buckling of these layers into the opening can occur (often with rockburst intensity and suddenness).

Deformation

The act of excavating an opening in rock is equivalent to removing the pressure that was acting to hold the remaining rocks in place. This removal of pressure is equivalent to applying on the boundary of the opening an increment of stress inward, as shown in Fig. 1(c), that causes the boundaries to deform. Drifts in the wall will be displaced towards the stope.

Elastic deflections in hard rock are small. However, the common experience of measured deflections increasing with time, without an increase in stope span, shows that failure takes time to develop.

In Fig. 1(d) dashed lines in the walls indicate potential zones of either tension or, at least, a reduction in compressive stress. Consequently, the rock structure will be loosened possibly permitting slabbing and sliding on fracture surfaces. The initial extent of such a zone of expansion, *excluding* the effects of any caving action, is between the quarter points of the span and extending for the elastic movement into the roof or walls for a distance equal to, at most, one quarter of the span. However, the high stresses may cause intense splitting (not uncommonly at spacings less than 0.1 in.) around the face and into the walls, or roof, in the direction of the stress trajectories, as shown in Fig. 1(d). In this way, the fractured dome around the zone of elastic expansion is formed. (If caving occurs, there is no limit to the height

[1]Modified version of report "ground control in deep mining" prepared for inclusion in the new Mining Engineers Handbook of the American Institute of Mining Engineers.

[2]Head, Mining Research Centre, Mines Branch, Department of Energy, Mines and Resources, Ottawa, Canada.

[3]Assistant to Chief Mines Engineer, International Nickel Company of Canada, Limited, Copper Cliff, Ontario, Canada.

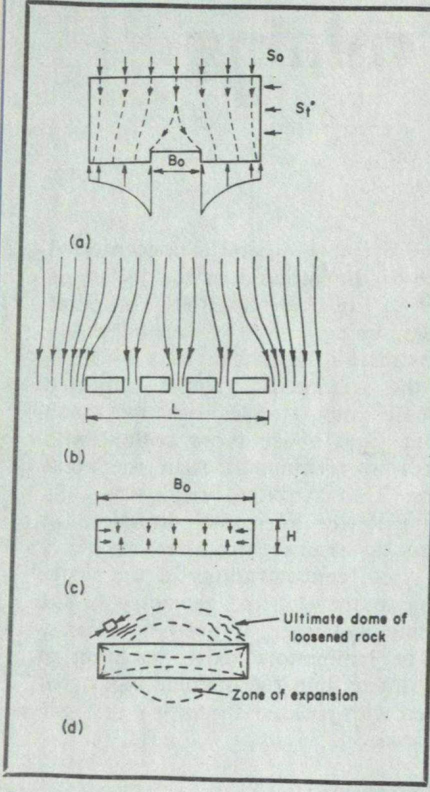

Fig. 1 Stress effects

of loosened or fractured ground.) The breakdown of the rock structure in such relaxed domes is the primary source of rockbursts in some areas. The enlargements that occur with increased stoping and the sudden joining of two adjacent domes, when either a pillar is removed or fails, or artificial supports are removed, can trigger particularly large rockbursts.

Strength

The stress-strain curve for a typical hard rock in compression is shown in Fig. 2(a). Zone I signifies the closing of fractures and joints; at the end of Zone II the rock starts to break down. There is some experimental evidence for believing that failure will occur at any stress level within Zone III, more time being required at the lower levels. Fig. 2(a) also shows the variation of noise, or microseism rate, with stress. For this typical hard rock the activity starts to increase conspicuously at the beginning of Zone III.

Fig. 2(a) also shows the unloading part of the stress-strain curve for a hard rock sample using a very stiff testing machine (one that is controlled so that it does not follow a failing sample) that is analogous to walls applying load on a pillar. It can be seen that, after the strength of the sample was exceeded, the rock remained intact because the rapid unloading spring action of a normal testing machine did not crush the sample to pieces. Cycling of stress at strains beyond the failure point showed that the material was still capable of sustaining some load. This experimental work throws some light on the behaviour of pillars that have been observed to be badly cracked but seemingly still provide some support for the walls (as well as pillars that have been the location of repeated bursts). Whether a rock mass will fail, either by yielding as indicated by such a stress-strain curve, by viscous deformation over time, or by exploding suddenly, depends on many factors. In any event, it is not yet possible to predict whether a certain rock mass will fail gently or with a bursting action on the basis of prior testing.

Another important aspect of strength is its dispersion around a mean value, which is common to all structural materials. Fig. 2(b) shows a frequency distribution curve, or a histogram, of the number of samples failing within each strength range. This makes it clear that it is impossible to predict failure but, at best, probabilities of failure, e.g. at a stress equal to the average strength 50% of the rock would fail, or at a stress of only 0.6 of the average strength approximately 15% of the rock would fail (depending on the actual shape of the bell-curve).

Confining pressure can greatly increase rock strength, which explains the experimentally determined variation (Fig. 2(c)) of strength, Q, with the ratio of breadth to height, B/H, of a pillar. The high strength for the high ratios results from the confinement occurring in the central part of a broad pillar, which has some implication with regard to the amount of energy that can be stored in such rock before it fails. It is known from experience in bursting rocks that below some minimum critical breadth, or B/H, failure will be by relatively gentle crushing. Furthermore, above some maximum critical size, (like 36,000 sf or a B/H= 50, in the South African gold mines), the pillar will not burst because it is akin to a solid abutment. Consequently, there can be a critical range within which the risk of rockbursts is great. Unfortunately, these minimum and maximum sizes can be determined only by local experience and even then only approximately.

Similar to broad pillars, some of the fractured rock in stope faces is held in place by the walls and hence builds up a back pressure that provides confinement for the inner rock. This confinement increases the strength of the inner rock and thus its ability to sustain very high stresses (and energy contents).

Rockbursts can be thought of as occurring in brittle rocks when gradual failure is prevented in the face, in pillars and in the walls. For example, the confinement occurring in a broad pillar inhibits early failure. Also, an abnormally strong section of rock that can not only store more than normal strain-energy but also attracts load from the adjacent working rock (like a hard inclusion) will not be failing while the adjacent sections are working and hence is a potential rockburst zone. De-stress blasting might be useful in such cases, but it is still not clear whether this technique always provides the assumed effective relief of stress. Another aspect affecting gradual failure is the advance rate, which can be too fast to permit the time consuming cracking and working at the face, characterized by Zone III in Fig. 2(a), that release stress concentrations. Of course, advance rates can also be so slow that disintegration and possibly bursting in the walls can ensue.

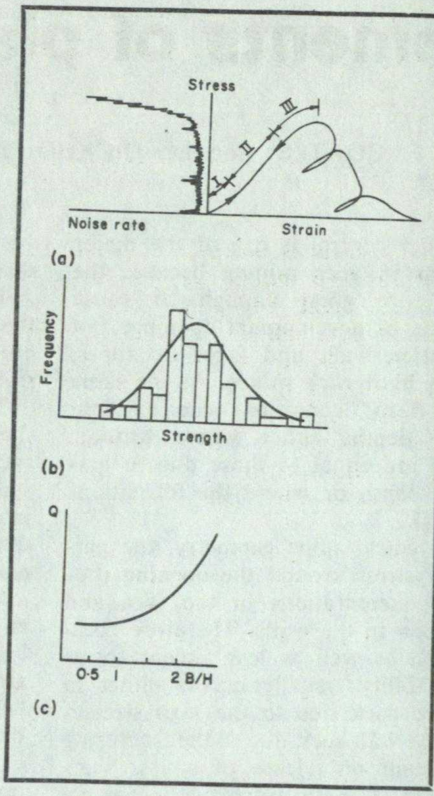

Fig. 2 Strength characteristics

Geology

Geological factors complicate most underground operations in a variety of ways that make each situation unique. In general, the rock mass will have a lower strength than the rock substance owing to the presence of the structural features (under favourable conditions it might be about ½).

In Fig. 3(a) the circular hole in a plate subjected to uniaxial pressure has been filled with a hard inclusion. Because the inclusion is more resistant to deformation than the plate material, it provides a greater reaction; or in other

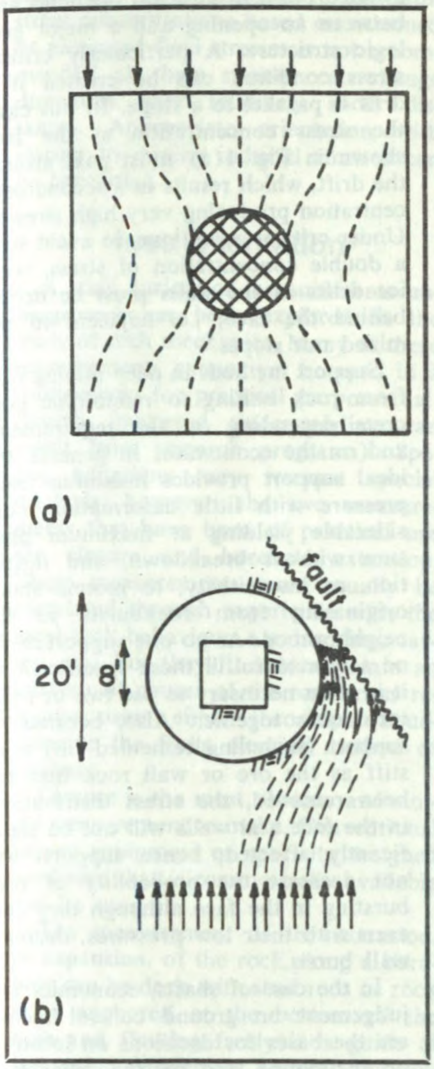

Fig. 3 Geological factors

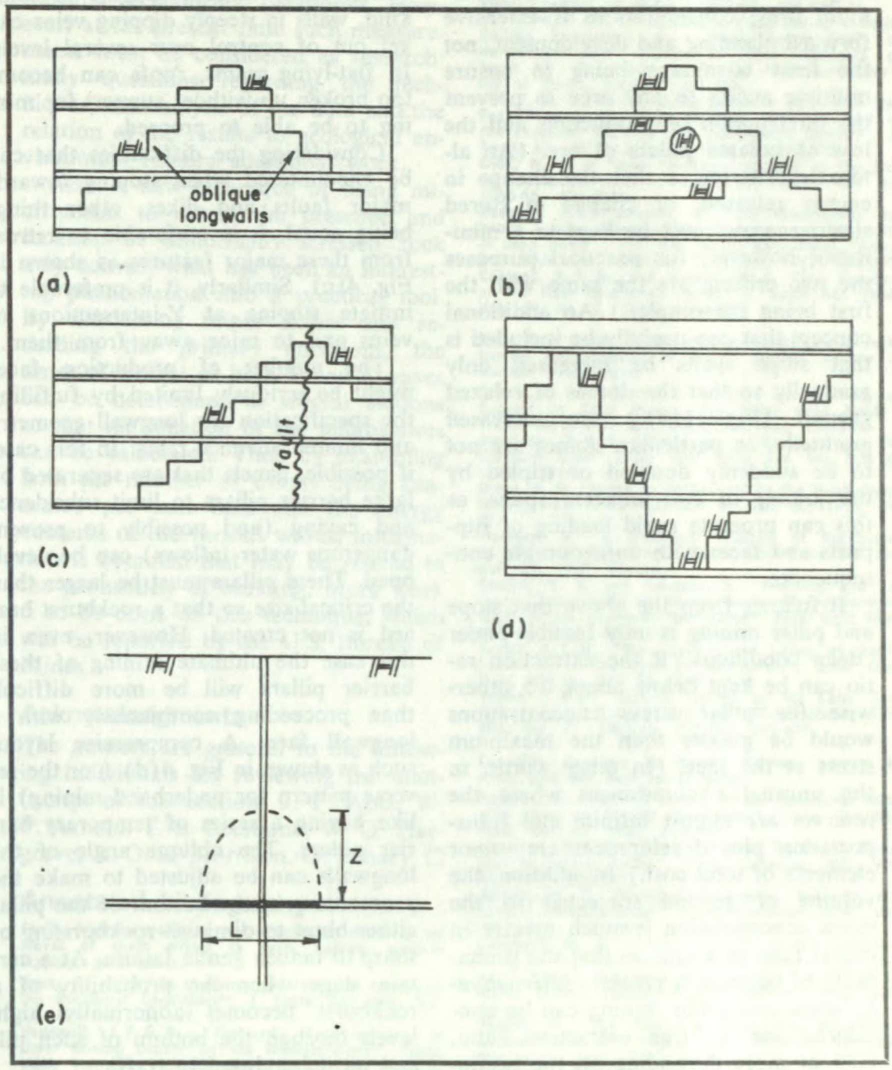

Fig. 4 Planning

words, it attracts load. Such an inclusion is analogous to dikes and layers that are more competent than the surrounding rock. Such dikes will attract more load and, as has been well documented, can be the source of more and bigger rockbursts than is normal for the formation.

If the hole in Fig. 3(a) had been filled with a softer material, the reverse pattern would have occurred, i.e. the inclusion would provide a lower reaction than the surrounding material making much of the stress pass around it somewhat similar to the pattern shown in Fig. 1(a). For this reason, mining towards a soft dike or fault zone can also be the cause of an increased frequency of rockbursts even in development headings where faces suddenly explode throwing rock and anything else in the way up the shaft or along the drift.

In Fig. 3(b) another mechanism is demonstrated. In this case, the lower shear resistance of a fault produced an abnormal concentration of stress and a rockburst around the 20-ft-diameter shaft that was being sunk on an 8-ft square pilot raise. The concentration of stress around the initial raise had not been great enough to cause trouble.

A few of the other geologically produced ground control problems can only be mentioned. Prominent joint sets permit wedges of wall rock to be easily detached and fall into stopes. Folds can be the source of zones that have been subjected to expansion and thus do not have the benefit of a certain amount of confinement, whereas other parts of the structures are subjected to additional compressive stresses making them more prone to failure when exposed.

Planning

Stoping

Investigation and testing, design, and fabrication are distinct and finite operations in the construction, manufacturing and aeronautical industries. Such concepts must be appropriately modified in mining so that they are considered as processes, not events, owing to the fact that mines are developed over a long time by excavating a series of openings in rock of varying properties and of great areal extent.

A system must be established for a continuous feedback circuit of: site investigation; planning of layouts and supports based on known and assumed conditions; calculations of measurable responses (rock deformations and stresses, pressure on supports) based on design assumptions; calculation of responses that would occur for possible deviations from the design assumptions (excessive water pressures); monitoring with instruments; additional investigations or testing indicated by the monitoring, or experience, as necessary; new plans; new calculations; monitoring — until the mine is closed down.

Planning layouts and production sequences must follow the rule that minimum stress concentrations be created at each stage and that no intermediate geometry be worse than the final condition. Stopes clearly create greater stresses in faces than development openings — the stresses increasing as the geometry becomes more elongated.

Implicit in the requirement for minimum stress concentrations is extensive forward planning and development, not the least advantage being to ensure multiple access to any area to prevent the interruption of production and the loss of isolated pillars of ore. (An alternate criterion is that the change in energy released, or change in stored strain energy, must be kept to a minimum; however, for practical purposes the two criteria are the same with the first being the simpler.) An additional concept that can usefully be included is that stope spans be increased only gradually so that the domes of relaxed ground (Fig. 1(d)) are increased gradually. In particular, domes are not to be suddenly doubled or tripled by the joining of two adjacent spans, as this can promote rapid loading of supports and faces with unfavourable consequences.

It follows from the above that stope and pillar mining is only feasible under "deep conditions" if the extraction ratio can be kept below about 0.5 otherwise the pillar stress concentrations would be greater than the maximum stress at the face. (In other words, in the unusual circumstances where the reserves are almost infinite and transportation plus development are minor elements of total cost.) In addition, the volume of ground subjected to the stress concentration is much greater in pillars than in a face so that the probability of bursting is greater. Alternatively, stope and pillar mining can be considered for a large extraction ratio, 0.85 or more depending on the economics, where it is judged that small pillars that crush gradually without bursting will provide the support required to control closure. The pillar sizes for this option must be smaller than the minimum critical size as described above with respect to bursting. Also, where zones of barren rock are to be left in place and stress conditions are critical, these 'horses' must be blasted if they fall within the range of critical pillar sizes.

Longwall geometry, whereby series of faces follow lines (Fig. 4(a)) that may be perpendicular or oblique to the strike, avoids excessive stress concentrations associated with the pillars, capes and peninsulas shown in Fig. 4(b). This approach is almost essential in deep mining because it is impossible to prevent caving (except where extraction ratios of less than 0.5 are feasible). Hence caving must be accepted and controlled, e.g. by using either fill support, packs, crushable pillars, or temporary unit supports in narrow veins (where ultimate complete closure can be accepted). Some support is normally required because rehabilitation of levels after a rockburst is thereby made easier. Also, without support of some kind, walls in steeply dipping veins can get out of control over several levels. In flat-lying seams, roofs can become too broken up without support for mining to be able to proceed.

Considering the difficulties that can be encountered when stoping towards major faults and dikes, other things being equal, it is preferable to retreat from these major features as shown in Fig. 4(c). Similarly, it is preferable to initiate stoping at Y-intersections of veins and to mine away from them.

The number of production faces might be seriously limited by fulfilling the specification for longwall geometry and limited advance rates. In this case, if possible, panels that are separated by large barrier pillars to limit subsidence and caving (and possibly to prevent dangerous water inflows) can be developed. These pillars must be larger than the critical size so that a rockburst hazard is not created. However, even in this case the ultimate mining of these barrier pillars will be more difficult than proceeding continuously with a longwall face. A compromise layout such as shown in Fig. 4(d) (or the reverse pattern for underhand mining) is like having a series of temporary barrier pillars. The oblique angle of the longwalls can be adjusted to make the promontory at the bottom of the pillar either blunt to diminish rockbursting or sharp to induce gentle failure. At a certain stage when the probability of a rockburst becomes abnormally high, levels through the bottom of such pillars must be closed to traffic.

The method of mining the barrier pillars must be based on the same rule that minimum stress concentrations are created throughout the operation, which usually means starting at the bottom of a pillar that extends down dip and mining upwards. Starting from one side of the pillar and proceeding to the other would produce a period of very high stress concentrations as the far side is approached.

For multi-vein orebodies, if one vein is mined out completely before the other, mining the hanging wall vein first is considered to be preferable, particularly if the orebody has a low dip so that the mining of the second vein would not be in caved ground.

Developing

For development openings such as drifts, crosscuts, levels, pumping stations, hoist rooms and haulage inclines, excessive stress concentrations are to be avoided. Hence openings must be arched, unless strong layering makes this inappropriate, to the shape that they would otherwise tend to work to. Appropriate pillars are to be left and acute angle intersections are to be avoided between adjacent openings and between an opening and a major geologic structure. A particularly critical stress condition can be created if a drift is parallel to a stope. In this case, the stress concentration at the face shown in Fig. 1(a) must pass around the drift, which results in a second concentration producing very high stresses. Under critical conditions, to avoid such a double concentration of stress, service drifts in the walls must be driven behind the face, i.e. adjacent to the mined out stopes.

Support methods in deep mining vary from rock bolting to monolithic concrete depending on the requirements and on the economics. In general, the ideal support provides maximum back pressure with little deformation, considerable yielding at maximum pressure without breakdown, and insulation, or insensitivity, to ground shock originating from rockbursts in the neighborhood. As no one support method seems to fulfill these specifications, it is often necessary to use two or more techniques together. Also, because no support (including cemented fill) is as stiff as the ore or wall rock that has been removed, the stress distributions in the face and walls will not be significantly affected; hence supports will not decrease the probability of rock bursting in the face although they can, even with their low pressures, decrease wall bursts.

In the case of shafts, economics and judgement on ground control factors are the bases for decisions on location, on inclination and on the pillar size. The size of shaft pillars must be determined substantially by judgement because no quantitative approach yet exists. In thin, flat orebodies where economics and ground control conditions favour their removal, it has been found easier to mine the shaft pillar before stoping adjacent ground. Indeed, it may be advisable to remove the pillar before the shaft is sunk, e.g. from inclines in the vein. The amount of ground affected by such removals will depend on the competence of the support and the width of stoping. In two cases using timber packs in narrow reefs, significant expansion of wall rocks occurred at a distance into the wall corresponding to a span ratio, Z/L, of close to 1 (see Fig. 4(e)). Because the exact value of Z/L is not predictable, the extent of cracking should be monitored, especially where flooding can occur. The shielding of other development openings can be achieved in the same way, i.e. by stoping or excavating a slot over important drifts and crosscuts (actually the slot should be normal to the maximum field stress, which is not necessarily vertical). The shaping (into ellipses) of shafts and other openings to minimize stress concentrations based on assumed or measured field stresses is not recommended as these stresses can change directions over relatively short distances. Appropriate orientation with respect to mining induced stresses can be beneficial.

Instrumentation

A wide variety of measurements underground has been explored in the study of rock mechanics. Many of these measurements cannot yet be used in a routine way for practical purposes (although improved understanding can result from research programs). Special situations may warrant special methods; however, the two measurements that have been of practical use are closure and borehole extension. These measurements must usually be correlated through experience with the local rock behaviour, e.g. bursting, caving, etc., but they fit naturally into an analysis of causes of the rock mass response using either theoretical equations or the finite element method of analysis.

Closure is the most obvious and simple measurement to make. Conventional survey equipment or specially designed hardware that is commercially available can be used.

The measurement of the extension, or expansion, of the rock along a borehole can be done with steel rods or rock bolts anchored at the bottom of the hole and floating freely in a guide at the collar. Movement between the collar of the hole and the anchor can be measured with micrometers or dial gauges. For longer holes, wires are used and multiple anchors can be installed at various locations along the borehole so that the expansion between sections can be obtained. Again, equipment can be developed at the mine, but commercially available hardware is usually preferable. The data provides good information for determining the reaction of pillars to pressure, the source of movement around development openings and the extent of cracking, or doming, in stope walls.

The other measurement that might be of some use in deep mining is that of the field stresses in the formation unaffected by mining. Such measurements might establish that residual stresses are higher than gravitational stresses in the area or in some restricted sections of the area, thus providing a warning of abnormal conditions. However, at the present time such measurements must be considered as research; many questions regarding the techniques, the analysis of the data and the relation of the results to geological environment require additional work.

Also, a new technique of using microseisms to detect the presence and location of abnormally stressed rock may convert what has been an interesting phenomenon into a practical tool. By eliminating minor noises and recording the primary emissions, the times of arrival of both P and S-waves can be determined at several stations so that the location of the microseisms can be calculated. Then by examining both the number of events being generated per unit time and the energy contents of the various waves, information is obtained that may be related to the probability of bursting. More work is to be done on this technique, which will be reported by the U.S. Bureau of Mines.

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UNDERGROUND MINING

Stope-and-Pillar Design for the Elliot Lake Uranium Mines

D. G. F. HEDLEY and F. GRANT,
Research Scientists,
Mining Research Centre, Mines Branch,
Department of Energy, Mines and Resources,
Elliot Lake Laboratory,
Elliot Lake, Ontario.

ABSTRACT

In 1958, the Ontario Department of Mines appointed a committee to study the accident situation and mining methods used in the uranium mines in the Elliot Lake district. This committee reviewed the existing knowledge on mine design, pillar support and roof spans and came to the conclusion that scientific knowledge had not yet reached the stage of producing rational design procedures which would reduce the dependence on trial-and-error methods.

Since that time, the Mining Research Centre has made numerous stress measurements in two mines to define the pre-mining and pillar stresses. A comprehensive series of tests has been done in the laboratory to measure the compressive strength of the rock; a geological survey of the region has been made to evaluate tectonic stresses; and a survey was done on mining conditions with regard to stable and unstable pillar configurations.

This information has been used to evaluate pillar design as effected by pillar width and height, depth below surface, extraction and pre-mining stress fields. The resultant design of pillar widths and corresponding extraction as the depth increased is in line with past and present experience in these mines. The information available on roof stability is still insufficient to design stable stope spans with the same degree of confidence; experience and trial-and-error methods are still required.

D. G. F. HEDLEY was born in northern England. He graduated with a B.Sc. (engineering) from the Royal School of Mines, Imperial College, London, in 1961. He continued his studies at the University of Newcastle upon Tyne, gaining his Ph.D. (rock mechanics) in 1965. The National Research Council of Canada granted Dr. Hedley a Post-Doctorate Fellowship with the Mines Branch from 1965 to 1967. Since 1967 he has been a research scientist at the Mining Research Centre, Mines Branch, of the Department of Energy, Mines and Resources.

F. GRANT was born and raised in western Canada. After army service, he attended the University of Alberta, graduating as a mining engineer. He worked in the coal mines of Alberta and B.C. before joining the Mines Branch. He has been involved in research in laboratories at Crow's Nest, B.C., Ottawa and Elliot Lake, Ontario. At present he is a research scientist at the Western Office of the Mining Research Centre in Calgary.

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INTRODUCTION

TWELVE URANIUM MINES in the Elliot Lake district were brought into production between 1954 and 1957, in response to the urgent world demand for this mineral. To fulfill their production contracts, it was necessary to mine large-scale excavations after a minimum amount of development. The orebodies are relatively flat and a system of stopes and pillars was used at all the mines. Mine design evolved from trial and error and from the experiences of other mines. As mining progressed, stope-and-pillar dimensions were modified to obtain greater extraction, and, if pillar or roof failures occurred, the extraction was reduced to provide more stable conditions.

In the early 1960's, the demand for uranium decreased, causing the closure of nine of the twelve mines. One of the existing mines was reopened in 1968, and a new mine was brought into production. At the same time, large-scale production at an existing mine was curtailed.

In 1958, the Ontario Department of Mines appointed a committee to study the accident situation and the mining methods in the Elliot Lake district. The report[1] of this committee reviewed the existing knowledge on mine design, pillar support and roof spans and concluded:

"there is much about pillar loading and pillar strength that we do not know. This gap, with present facilities, can be greatly narrowed for the stronger rock types. Stress measurements as a practice are in their infancy but making progress. A better knowledge of stress distribution in mines is a prerequisite for improved mine designs . ."; and ". . . . at the present time, scientific knowledge can be applied to the problems of ground control only in a qualitative manner. Quantitatively the answers must still be confirmed by actual practice. In many cases, this will probably continue to be so, but in others, a better organization and application of data that has been and can be made available to the industry would greatly reduce dependence on trial and error methods."

Since the writing of this report, the Mining Research Centre has made numerous measurements in two uranium mines to try to define the pre-mining and pillar stresses[2,3,4,5]. A comprehensive series of tests has been done in the laboratory to measure the compressive strength of the rock[6]. A geological survey of the region and individual mines has been made to evaluate tectonic stresses[6,7]. Further experience has been gained by the operating mines as stope and pillar dimensions have been modified to suit local conditions and as their workings extended deeper.

At this time, it seems worthwhile to re-evaluate the bank of data and experience available on the Elliot Lake mines and to assess, in engineering terms, the measurements which have been made, the trial-and-error experience which has been gained, and the pillar and roof failures which have occurred. A similar evaluation has recently been made of the bord and pillar workings in the South African coal mines[8, 9]. Occurrences of pillar failure were documented and analyzed in terms of pillar load and strength to define acceptable pillar design.

A questionnaire was sent to each of the operating mines, and to persons who had experience in the closed-down mines. Information was requested on geological conditions, methods of mining, mining dimensions, artificial support, and on cases of pillar and of roof failure. Comprehensive sets of data were obtained from the operating mines and from some of the closed-down mines. Detailed information on ground control problems in some mines which have been closed for almost 10 years could not be obtained.

GEOLOGICAL ENVIRONMENT

The Elliot Lake area is about 20 miles north of Lake Huron and about halfway between Sudbury and Sault Ste. Marie. The area is characterized by Proterozoic sediments (up to 7000 ft thick) overlying Archean granites, greenstones and greywackes. The uranium-bearing conglomerates, 6 to 30 ft thick, are situated close to the base of the Matinenda formation, which is in contact with the Archean basement. Various uranium-bearing reefs occur separated by 10- to 100-ft-thick quartzite beds. Basement rocks and the sedimentary sequence have been intruded by numerous diabase dikes and sills.

The geological structure can be described as a broad syncline with an east-west axis plunging about 5 degrees west, dissected by northwest-trending faults. The last discernible tectonic event involved the development of east-west-trending and steeply dipping joints which have been caused by a maximum compressive stress acting in an east-west direction[6, 7].

In-situ measurements of pre-mining stress have been taken in two mines, one located on the southern limb of the syncline and the other on the northern limb. It was found that the vertical stresses were compatible with the weight of the overburden. Horizontal stresses were in the order of twice the magnitude of the vertical stress. In the north-south direction, the mean stresses at various measuring sites varied between 2500 and 3200 psi and, in the east-west direction, between 3000 and 5300 psi[4, 5]. This confirmed the deductions of the geological survey on the pre-mining stress field.

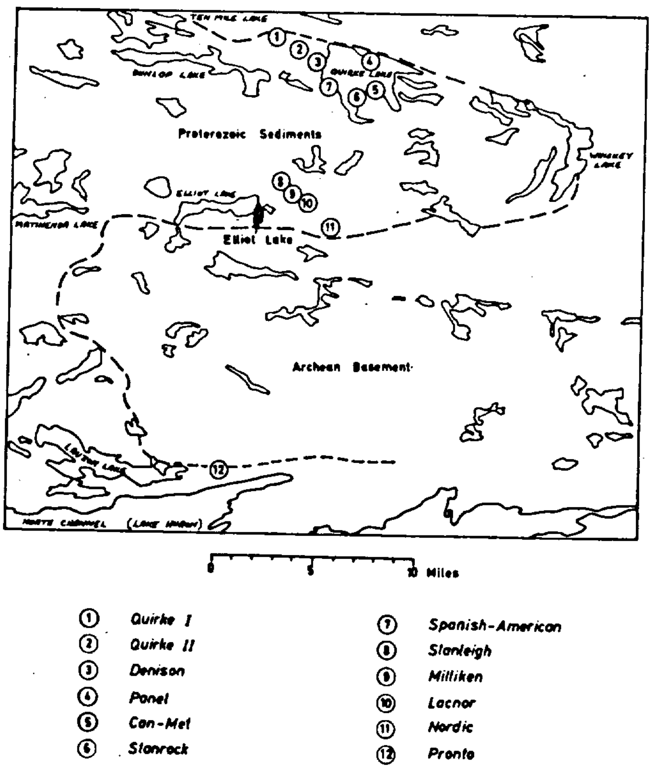

FIGURE 1 — Location of uranium mines in the Elliot Lake area.

MINING METHODS

Figure 1 shows the location of the twelve mines in the Elliot Lake area. There are seven mines situated on the northern limb of the syncline (Quirke I, Quirke II, Denison, Panel, Can-Met, Stanrock, Spanish-American) and four mines on the southern limb (Stanleigh, Milliken, Lacnor, Nordic). Another mine (Pronto) is situated about 20 miles to the south. Because the orebody is relatively flat, all the mines used a room- or stope-and-pillar method of extraction. To get into production as fast as possible, most mines did their development within the orebody, advanced the workings away from the shafts to the property boundaries and used trackless equipment.

Development headings were driven along strike and the rooms or stopes were mined on dip. Where the orebody was over 10 ft thick and its dip less than about 15 degrees, trackless equipment was used for drilling, loading and transporting. Approximately square pillars were left on regular or random patterns. Where the orebody was less than 10 ft thick or the dip was steeper than 15 degrees, scrapers were used. With this system, narrow pillars about 250 ft long were left on dip with sill pillars on strike at the ends of the stopes. The present operating mines all use the scraper stoping system. There are a number of papers in the technical literature describing the mining methods used by different mines[10, 11, 12, 13].

In many mines, more than one ore zone exists. If two zones are close together they are mined as one, otherwise they are mined separately. In the latter case, it is tried to keep the pillars in both zones directly over one another. Where conservative stope and pillar dimensions were originally used the mines have gone back and removed or decreased the size of some of the pillars.

Table 1 lists the pertinent information on mining dimensions at the twelve mines. Depth below surface ranges from outcrop to 3500 ft. The dip of the orebody can vary considerably due to local rolls, but it averages just over 20 degrees for the northern limb of the syncline and just under 20 degrees for the southern limb and less than 10 degrees at depths greater than 3000 ft. The mining height depends on the number of ore horizons being mined; the average for single reefs ranges from 6 to 15 ft. The extent of the mining area is quite extensive for those mines longest in operation and, in most cases, the lateral extent of workings is greater than the depth.

Dimensions of stopes and pillars have varied considerably as to whether a room-and-square pillar or a stope-and-rib pillar system was used. Present mining practice is fairly uniform, with 65-ft stope spans extending about 250 ft on dip. Pillar widths and heights are usually maintained at a 1:1 ratio. Extraction has ranged from 60 to 90 per cent, with in general the greatest extraction being obtained in the shallowest mines and the least extraction in the deepest mines.

TABLE 1 — Mining Parameters in the Elliot Lake Mines

Mine	A	B	C	D	E	F
Mining Method	Stope & Pillar	Stope & Pillar	Stope & Pillar	Stope & Pillar	Stope & Pillar	Stope & Pillar
Depth, ft	0 — 800	20 — 1050	300 — 1200	50 — 1450	1000 — 1600	2200 — 2600
Dip range°	15 — 50	15 — 40	5 — 40	10 — 20	15 — 40	14 — 20
average°	20	20	12	17	28	17
Ore horizons	1	2	1	2 separately	1	2 separately
Mining height, ft	7.5	5-12, ave. 9	14	7 upper 7-12 lower	7-9.5, ave. 8	5-12 upper 6-12 lower
Area Mined Strike × Dip, ft	4000 × 2500	7000 × 2000	3500 × 1500	6000 × 4000	1300 × 1400	3500 × 1400
Stope Dimensions						
length, ft	irregular	150 — 300	150 — 200	200 — 400	220 — 360	150 — 300
span, ft		65	55 — 70	65	65	50
Rib pillars						
length, ft		150 — 300	150 — 200	200 — 400	220 — 360	150 — 300
width, ft	irregular	10	10 — 15	10	10	10
Sill pillar width ft		15		15	10	15
Extraction %	90	85	85	85	85	75

Mine	G	H	I	J	K	L
Mining Method	Stope & Pillar	Room & Pillar Stope & Pillar	Room & Pillar	Room & Pillar Stope & Pillar	Room & Pillar Stope & Pillar	Stope & Pillar
Depth, ft	2600 — 3000	600 — 3000	1700 — 3000	3000	3000 — 2500	3000 — 3200
Dip range°	11 — 17	0 — 60		5 — 15		
average°	12	18	12		8	17
Ore horizons	2 separately	2 together	1	2	2 separately	1
Mining height, ft	7-12 upper 7 lower	5-32, ave 15	14	5.5-8, ave 6	10 upper 10 lower	10
Area Mined Strike × Dip, ft	5000 × 3000	15000× 10000	1700 × 2000	7000 × 2500	3000 × 2500	2000 × 500
Stope Dimensions					Rooms Stopes	
length, ft	200 — 300	200 — 250	rooms	200 — 250	irregular 200	200
span, ft	30-100, ave 60	25-110, mainly 65	20-25 wide	35	50	50
Rib pillar						
length, ft	200 — 300	40 200-250 × and ×	25-30	200 — 250	200	150 — 200
width, ft	10 — 20	20 20-25	20	15	irregular 10	20 ave.
Sill pillar width, ft		15 — 20		20 — 25		25
Extraction %	65 — 85	65	70	60 — 70	70	65

The following is an example of the trial-and-error process of mine design which one mine went through to arrive at acceptable stope and pillar dimensions.

Its orebody is between 2580 and 3000 ft below the surface and dipping at 11 to 17 degrees. The original design permitted 68 per cent extraction in 60-ft-wide stopes between triangular rib pillars on dip. While the first stope was being mined, deteriorating roof conditions were experienced when the span reached 30 ft. The layout was changed to 30-ft-wide stopes and 10-ft-wide pillars, representing 75 per cent extraction.

It was found that the roof conditions in the first stope were an exception rather than the rule, and the stope spans were increased to 40 ft to permit 82 per cent extraction. Because no ground control problems were encountered, the stope widths were increased to their original 50 ft between 10-ft-wide pillars for an extraction of 85 per cent. In some places, ore about 20 ft below the main orebody was mined. The top section was mined 7 to 12 ft high, the bottom section 7 ft high, leaving an intervening bridge 15 to 25 ft thick.

To obtain 82 per cent extraction and less development, the stope widths were increased to 100 ft, leaving 20-ft-wide rib pillars and small pillars on strike at the mid-length of each stope. At this point, pillar failures occurred in some of the older workings, where both the lower and upper horizons had been mined, some pillar recovery had been done and a fault passed through the area. The haulage drifts, about 30 ft below the sill pillars, started to deteriorate. This was alleviated by removing the sill pillars and, hence, the stress concentration. A survey of the existing pillars showed that those 10 ft wide were deteriorating or had failed. Also, the backs of some stopes had caved near fault zones. All faults in the mine were then surveyed so that additional pillar support could be left where required. The stope layout was revised to give 75 per cent extraction — two 70-ft-wide stopes with a 15-ft-wide centre pillar between them and 20-ft-wide rib pillars on the outside. Also the sill pillars directly above the haulage drifts were systematically removed at the end of the stoping cycle.

Pillar and roof failures continued to occur in the older sections of the mine. A further revision was made to the stope layout. A stope 50 ft wide was mined along strike directly above the haulage drift. With this completed, the remainder of the panel was mined in two 60-ft-wide stopes for an extraction of 70 per cent. Pillar widths were standardized at 20 ft and extra pillars along strike were left at the junction of the up-dip and down-dip mining stope. This design appeared to give stable conditions.

In review it seems that pillars 10 ft wide started to fail where the extraction was greater than 80 per cent, and especially in areas where two horizons were mined. Roof caving occurred if the span was 70 ft or more, especially where geological-weakness planes existed. Sill pillars concentrate stress and troublesome conditions on haulage drifts. Stopes 60 ft wide and pillars 20 ft wide with an extraction of 70 per cent appear to give stable conditions.

PILLAR STABILITY

Pillar stability can best be expressed in terms of a safety factor which is defined as

$$\text{Safety Factor} = \frac{\text{Strength of a pillar}}{\text{Stress applied to a pillar}} \quad (1)$$

A safety factor greater than one represents stability; less than one, instability. It is unlikely that an exact value of the safety factor can be calculated because the information on pillar dimensions, extraction ratios and pre-mining stresses is not very accurate. Consequently, pillar stresses and strength can only be estimated and then examined to see if they are in agreement with past and present mining conditions.

Estimation of Pillar Stress

The stress acting on a pillar has been shown[14] to depend on pre-mining field stresses, extraction ratio, location of the pillar, width and height of the pillar, and the physical properties of the rock. Of these factors, only the pre-mining field stresses and the extraction ratio have a major effect on the pillar stress. Hence, provided the mined area is large enough, a simplified equation can be used to relate pillar stress to pre-mining stress and to extraction ratio.

$$\sigma p = \frac{So}{1 - R} \quad (2)$$

where σp = pillar stress,
So = pre-mining stress normal to orebody,
R = extraction ratio.

For inclined workings, the normal stress 'So' is a combination of the components, vertical stress 'Sv' and the horizontal stress 'Sh';

$$So = Sv \cos^2\alpha + Sh \sin^2\alpha \quad (3)$$

where α = dip of orebody.

The vertical stress (Sv) can be assumed to be the weight of the overlying strata, which increases at about 1.1 psi/foot depth (density of rock 160 lb/cu.ft);

$$Sv = 1.1\,D \quad (4)$$

where D = depth from surface (ft).

The horizontal stress is more difficult to estimate, especially if the area has been subjected to tectonic stresses. Measurements in two mines indicate a horizontal stress perpendicular to strike (north-south direction) of about 3000 psi[4,5]. Consequently, the normal stress 'So' can be expressed as

$$So = 1.1\,D \cos^2\alpha + 3000 \sin^2\alpha \quad (5)$$

and the pillar stress by

$$\sigma p = \frac{1.1\,D \cos^2\alpha + 3000 \sin^2\alpha}{1 - R} \quad (6)$$

Figure 2 shows a nomograph relating pillar stress to depth 'D', dip 'α' and extraction ratio 'R'. An example of how to calculate pillar stress is outlined on the nomograph: at a depth of 2000 ft, for an orebody dipping at 20 degrees, the pre-mining normal stress is 2300 psi and, for an extraction of 80 per cent, the pillar stress would be 11,400 psi. It can be seen that, where the dip is less than 20 degrees, the horizontal stress has little effect on the pillar stress.

This method of estimating pillar stress can be compared with *in-situ* measurements of pillar stress made at two mines[2,3]. However, the measurements themselves cannot be taken as accurate values and can vary by ± 50 per cent. Table 2 shows the comparison for five measuring sites ranging in depth from 920 to 1180 ft. The stress normal to the orebody was calculated for both the measured and estimated stresses. In most cases, there is good agreement between the average measured and estimated stresses. Any discrepancy is covered by the range of the mea-

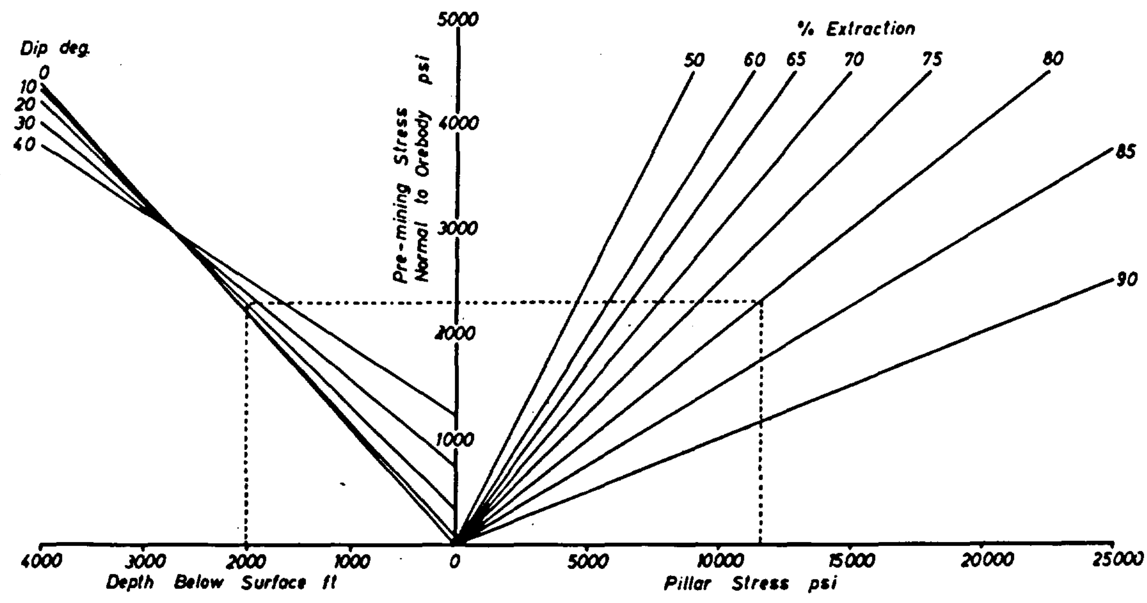

FIGURE 2 — Determination of pillar stress.

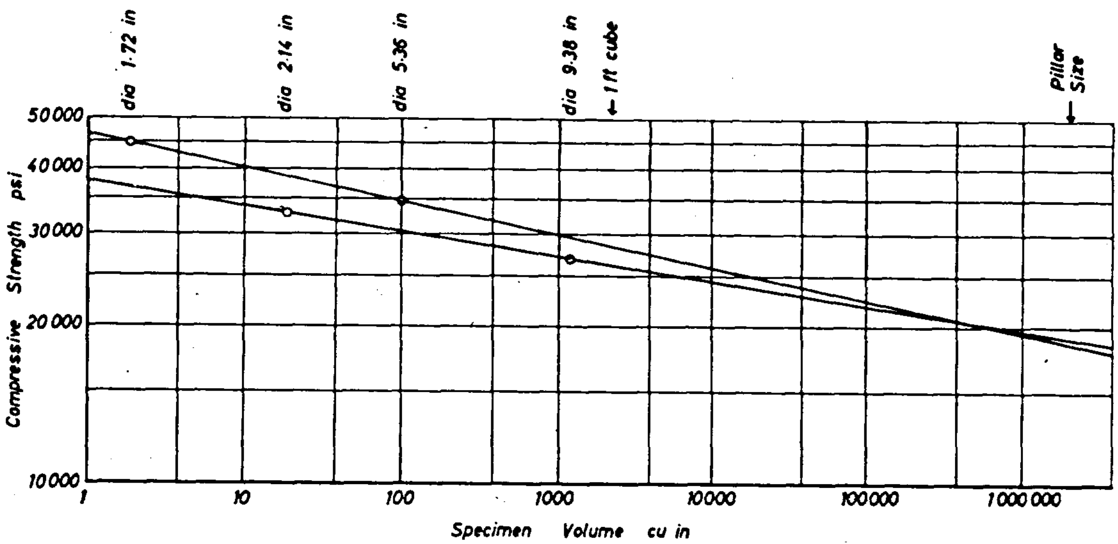

FIGURE 3 — Effect of specimen size on compressive strength (after Kostak).

TABLE 2 — Comparison of Measured and Estimated Pillar Stresses

Depth ft	Extraction %	Measured Stress, psi		Estimated Stress, psi	
		Range	Average	Average	+ 5% Extraction
920	85	5500 — 13700	8700	8000	6000 — 12000
970	85	6300 — 12300	8100	8300	6200 — 12500
1050	85	7900 — 10600	9600	8900	6600 — 13300
1050	40	1500 — 3200	2400	2000	1800 — 2200
1180	85	4800 — 6800	5600	9700	7300 — 14600

sured and estimated stresses, taking into account that the local extraction can easily vary by ± 5 per cent. An exception is the site, 1180 ft deep, where the measured stress is only 60 per cent of the estimated stress and the range of both measured and estimated stresses does not overlap. This discrepancy may be due to varying field stresses and the site being near the abutment.

Estimation of Pillar Strength

The strength of a pillar cannot be measured directly but must be estimated from small-scale laboratory tests and from whatever information is available from actual *in-situ* pillar failures. Laboratory-determined strengths are for specimens having a volume of about 10 cu.in. whereas *in-situ* mine pillars have a volume of about 2 million cu.in. Consequently, there can be large inaccuracies in extrapolating the strength of laboratory specimens to mine pillars.

Over 400 specimens, ranging in diameter from 1.72 in. to 9.38 in., have been tested in the laboratory[8]. The relation between specimen size and strength is shown in Figure 3. It can be seen that as size increases the strength progressively decreases. These laboratory-determined strengths have been extrapolated to usual pillar dimensions and indicate a pillar strength of about 19,000 psi.

TABLE 3 — Dimensions and Estimated Stress and Strength of Pillars

Depth ft	Dip deg.	Extraction %	Pillar Width ft	Pillar Height ft	Estimated Pillar Stress psi	Estimated Pillar Strength psi
Stable Pillars						
500	17	85	10	10	5000	14600
700	17	85	10	10	6400	14600
800	26	65	20	18	3800	13300
850	20	85	10	10	7600	14600
950	11	85	10	10	7500	14600
1000	22	65	20	18	4000	13300
1050	15	85	10	10	8500	14600
1200	18	85	10	10	9400	14600
1300	20	65	20	20	4600	12300
1600	20	60	20	18	4800	13300
1600	20	65	18	18	5400	12600
1600	22	75	20	14	7600	16000
1700	22	65	40	20	5800	17400
1700	22	60	22	20	5000	12900
1700	12	75	20	14	7600	16000
1800	5	75	20	14	8000	16000
1900	23	65	19	18	6400	13000
2200	25	65	20	20	7200	12300
2400	11	65	20	8	7600	24400
2500	9	65	20	8	7900	24400
2700	13	65	20	8	8600	24400
2900	12	70	15	9	10500	19400
2900	12	75	20	9	12600	22400
Partially Failed Pillars						
1400	20	85	10	10	11400	14600
2400	18	80	10	9	13400	15800
Crushed Pillars						
2800	12	80	10	9	15200	15800
2900	12	80	10	9	15700	15800
3400	5	80	15	10	18500	17900

Information was collected on both stable and unstable pillar dimensions in the mines. The depth, dip, extraction, and pillar width and height are given in Table 3. There are twenty-three cases of stable pillar dimensions, two cases where partial failure of the pillars occurred and three cases of complete pillar crushing. The information on complete pillar crushing was obtained second-hand because it happened in mines which are closed. The stresses acting on these stable and unstable pillars were estimated, using equation 6, and listed in Table 3.

The relationship between pillar dimensions and strength is usually expressed in the form[8,15,16,17]:

$$Qu = K \frac{w^a}{h^b} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (7)$$

where Qu = pillar strength, psi,
w = pillar width, ft,
h = pillar height, ft,
K = strength of 1 ft cube, psi,
a and b = constants.

This equation refers to square pillars, whereas those in the uranium mines are usually long and narrow. However, it is considered that the strength of such a pillar will not be very much greater than that of a square pillar of width equalling the minimum width of the long pillar.

The strength of a 1-ft cube is obtained from the laboratory tests and, as shown in Figure 3, is 26,000 psi (equals K). The values of constants 'a' and 'b' quoted in the literature are:

Greenwald *et al.*(15)	a = 0.5	b = 0.83
Steart(14) and Holland *et al.*(17)	a = 0.5	b = 1.0
Salamon *et al.*(8)	a = 0.46	b = 0.66

The value for 'a' is relatively constant at 0.5, whereas that of 'b' varies over a larger range.

The estimated stress (and hence pillar strength) acting on the three crushed pillars was substituted into equation 7, along with the respective dimensions of the pillars, a 'K' value of 26000 psi and 'a' equal to 0.5. Three values for 'b' were calculated ranging from 0.736 to 0.768, with a mean value of 0.75. Consequently, pillar strength can be related to pillar width and height by

$$Qu = 26000 \frac{w^{.5}}{h^{.75}} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (8)$$

This equation was used to calculate the strengths of all the pillars listed in Table 3.

Estimated pillar stress is plotted against pillar strength for all the pillar cases in Figure 4; in addition, lines representing safety factors from 1.0 to 3.5 are drawn. It can be seen that the results for the crushed, partially failed and stable pillars are compatible with the observations. The safety factors of crushed pillars are grouped around 1.0, those of partially failed pillars lie between 1.0 and 1.3, and those of the stable pillars all exceed 1.5.

Pillar Design

The stress acting on pillars and their strength have now been defined in terms of pre-mining vertical and horizontal stresses, dip of orebody, extraction ratio, and pillar width and height. Consequently, the safety factor can also be described in these terms by substituting equations 6 and 8 into equation 1.

$$\text{Safety factor} = \frac{\dfrac{26000\, w^{.5}}{h^{.75}}}{\dfrac{1.1\, D \cos^2\alpha + 3000 \sin^2\alpha}{1 - R}} \quad \ldots\ldots\ldots \quad (9)$$

This equation can now be used to determine the minimum pillar width and corresponding extraction as depth increases. Two examples have been calculated using the following conditions:

(1) a safety factor of 1.5 because this value appears to represent the dividing line between uncertain and stable conditions;
(2) a 20-degree dip of the orebody;
(3) pillar heights of 10 ft and 20 ft which mark the range of current mining heights; and
(4) a stope length of 250 ft and rib pillars spaced along strike at 75-ft centres, an average stope configuration.

The extraction ratio can be expressed in terms of the stope dimensions and pillar width, assuming that the sill pillar is of the same width as the rib pillars —

$$R = \frac{(250 - w)\,(75 - w)}{250 \times 75} \quad \ldots\ldots\ldots \quad (10)$$

Figure 5 shows the variation in pillar width and the corresponding extraction as depth increases to 3500 ft for 10- and 20-ft pillar heights. For instance, at a depth of 1000 ft, a 10-ft-high pillar requires a pillar width of 8.6 ft for an extraction of 85 per cent. A 20-ft-high pillar requires 12.3-ft width for a 79 per cent extraction. At the 3000-ft depth, the width of a 10-ft-high pillar has doubled to 16 ft and the extraction has decreased to 73 per cent, whereas the width of a 20-ft-high pillar has increased to 23 ft and the extraction decreased to 63 per cent. These pillar widths and extractions do not contradict those which have evolved through experience in the mines.

The widths of the 10-ft-high pillars are more reliable because information exists on pillars of this height which have crushed. No such information exists for pillars 20 ft high, therefore the variation in pillar strength with height cannot be checked.

This is about as far as the engineering analysis on pillar design can be taken. The quality of the information and limitations of the relationships prevent a more exact solution. Extraction could possibly be further increased by extracting the pillars on a retreating system. However, the resultant transfer of load and effect on pillar and roof stability is unknown; therefore, the experience of the mining engineer must be relied on.

ROOF STABILITY

The information available on roof stability is limited and the influence of various parameters is only vaguely understood. Stope span is very important, but the importance of roof thickness is not known. For instance, the stability of an 8-ft-thick bed spanning 65 ft would differ from that of a 30-ft-thick bed. Structural geology such as faults and joints can so enormously affect stability that the roof of a 20-ft-wide roadway can fail, whereas in another location an opening 300 ft square has been mined without roof collapse.

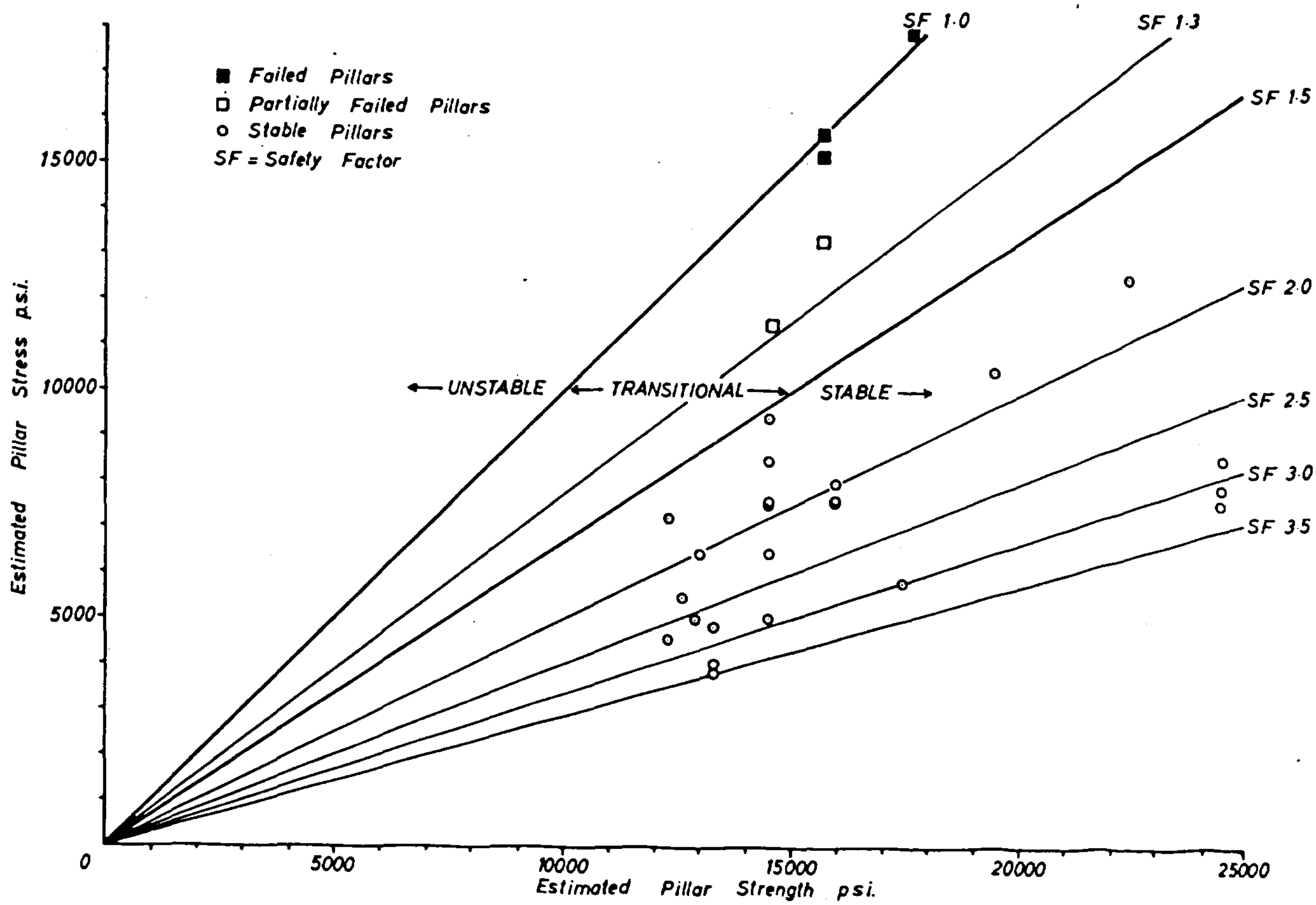

FIGURE 4 — Estimated pillar stresses and strengths.

At present, the best indication on safe mining spans is obtained by examining the stope spans which have evolved over the years in the mines. Initial mining produced stope and room spans ranging from 20 to 110 ft, but, by the mid 1960's, most stopes were from 50 to 65 ft wide. One mine has documented 18 cases of roof falls ranging in thickness from 2 to 17 ft. Of these, 17 could be explained by thrust faults in the immediate roof, by converging fault or slip planes or by prominent bedding planes in the roof. The one remaining fall extended 65 by 65 ft and arched up to 12 ft into the roof.

All the roof falls can possibly be classified as local, and there has been no indication of deep-seated roof failure. All the mines use rock bolts as a means of artificially supporting the roof.

SUMMARY

There has been a considerable advance in the knowledge available on mine design since the Special Committee on 'Mining Practice in the Elliot Lake Area' was published in 1961. Using information on the dimensions of stable and failed pillars in the mines in conjunction with laboratory strength determinations and *in-situ* stress measurements has resulted in a more rational and engineering approach to pillar design. The analysis on pillar stability showed that there was no contradiction between the calculated stability and that observed underground. In other words, the pillars that were supposed to be stable were stable and those which had totally or partially failed were estimated as such. This gives confidence

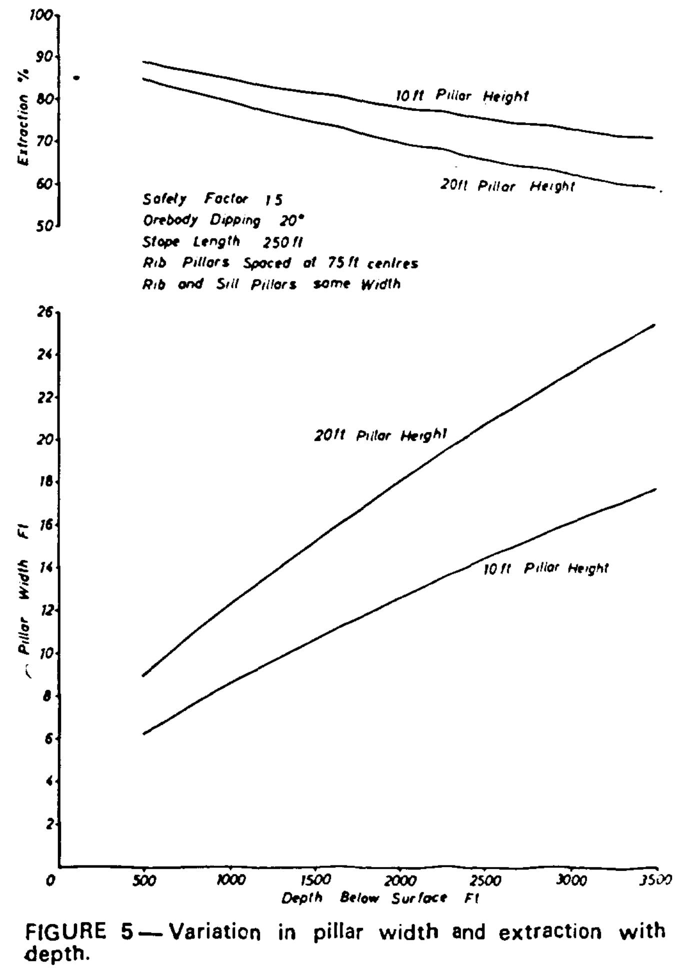

FIGURE 5 — Variation in pillar width and extraction with depth.

in applying this method of analysis to determine the minimum pillar widths and maximum extraction as the mines progress deeper.

Much more information is required on roof conditions with regard to documentation of existing falls, geological conditions and *in-situ* measurements before roof stability can be analysed in a similar manner as pillar stability.

ACKNOWLEDGMENTS

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